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METALLURGICAL TREATMENT OF ZINC-RETORT RESIDUES

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ILLINOIS

BY

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THE METALLURGICAL TREATMENT OF ZINC-RETORT RESIDUES

By B. M. O'Harra

INTRODUCTION

The residues resulting from the retort distillation of zinc ores have 25 to 75 per cent of the weight of the original ore and contain 5 to 15 per cent zinc. They consist of unconsumed coal or coke and the constituents of the ore that are not volatilized during the distillation of the zinc.

During the years 1918, 1919, and 1920 retort smelters in the United States produced an average of 443,000 tons of distilled primary zinc,[1] as distinguished from electrolytic and redistilled secondary zinc. This average excludes the exceptionally large production of 1917 and the abnormally low production of 1921. In round numbers, 450,000 tons may be considered the present normal production of zinc by retort smelters. On the assumption that about 1 ton of zinc is produced from 2 tons of concentrate, there must be normally about 900,000 tons of concentrate treated annually in retort smelting plants. If the weight of retort residues is estimated to be 40 per cent of the weight of the concentrates smelted and is assumed to have an average zinc content of 8 per cent, we can calculate that approximately 360,000 tons of residues, containing about 28,800 tons of zinc, are produced annually in the United States. This estimate is very crude, but as most zinc smelters neither weigh their residues nor sample them systematically an accurate estimate is not possible. Besides the current production of residues there are at many smelter sites large accumulations from past operations available for treatment whenever treatment becomes profitable.

In addition to zinc, the residues contain much unburned coal, carbonized or converted to coke during the distillation process; residues from most western zinc ores also contain lead and silver.

The treatment of these residues for the recovery of the contained coke and metal values presents a fertile field for investigation and has received the attention of many metallurgists, as is manifest from the number of methods that have been tried and of processes that have been patented.

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[1] Siebenthal, C. E., and Stoll, A., Zinc in 1921: U. S. Geol. Survey Mineral Resources, 1921, pt. 1, 1922, pp. 21–33.

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During the year 1922 the Bureau of Mines received so many inquiries concerning the possibility of profitably treating retort residues that an investigation of residue utilization was deemed advisable. As a first step a survey was made of the processes already tried and of those now in use. The treatment of broken retorts and condensers was included in the survey as a closely related subject. This survey showed that there was little occasion for further experiments of a general nature, but that the management of each individual zinc-smelting plant must decide for itself, according to local conditions, the feasibility of and the best methods for treating residues profitably. In order that the information collected in the course of the bureau's survey may be available to those interested in the subject, it is summarized in the following pages.

ACKNOWLEDGMENT

The author wishes to acknowledge the kindness of the zinc companies that have cooperated by giving information concerning the treatment of residues at their smelters and by allowing the writer to visit their plants.

TYPICAL ANALYSES OF RETORT RESIDUES

Some analyses of zinc retort residues which contained as low as 2 or 3 per cent of zinc have been reported, but the usual zinc content is considerably greater. The residues from calamine ores and low-grade ores have a lower zinc assay than those from high-grade sulphide ores. According to reported analyses the residues produced by European smelters contain generally less zinc than those produced in the United States. In this country the residues made by those plants which use a fairly high proportion of reduction fuel in the charge mixture contain 5 to 10 per cent of zinc—the residues from furnaces fired with producer gas running lower as a rule than those from furnaces fired with natural gas. In recent years some smelters have greatly reduced the proportion of reduction fuel and increased the weight of charge to the retorts. At these plants the residues tend to retain a higher percentage of zinc, but they weigh less per ton of ore charged.

The zinc content of the residue differs greatly in different retorts and in different parts of the same retort. In some tests made by Mühlhauser[2] it was found that the zinc content of the residue from retorts in different parts of the furnace ranged from 2.9 to 14.5 per cent—the general average of that from all retorts being about 7 per cent. In different parts of individual retorts the zinc assay ranged from 2 to 7 per cent except in the front 8 inches, where it increased rapidly—the residue in the front 2½ inches of the retort assaying 50 to 60 per cent zinc. About 50 per cent of the total zinc remaining in the residue was within 8 inches of the mouth of the retort.

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[2] Mühlhauser, Otto, Über den Zinkgehalt der Räumasche und dessen Bedeutung für die Muffelhaltbarkeit: Met. und Erz, vol. 15, Aug. 8, 1918, pp. 259–265.

The zinc is present partly as zinc sulphide and partly as zinc oxide and silicate. According to Schuchard,[3] from 12 to 43 per cent of the zinc is present as zinc sulphide, an average of about 80 analyses giving 31 per cent; this proportion seems to be independent of the total zinc content of the residues—at least of those containing between 0.15 and 7.0 per cent. Prost found that in a residue containing 4.76 per cent zinc, 2.36 per cent was in the form of sulphide.[4] Table 1 gives some typical analyses of residues from retort smelters in the United States.

TABLE 1.—*Typical analyses of retort residues*

	A	B	C	D	E	F	G	H
	Per cent	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*
Zn	8.45	6–8	5.8	11.6	10.2	7.5	5.5	14.5
Pb	4.75	0.5	3.9	0.11	4.7	9.0	3.4	2.2
Cu	2.08		0.35		0.6	0.2		1.0
Fe			11.8	5.55	7.2	11.0	29.7	29.9
Insol			23.6	31.5	30.0	27.5		
SiO_2	20.85						11.0	10.5
CaO	1.35		0.8	1.20				
S	3.38			2.33			3.1	4.8
C		42.0	32.4	41.0	35.0	30.0		
Mn	0.78				0.7	0.4	0.84	
	Ounces per ton		*Ounces per ton*		*Ounces per ton*	*Ounces per ton*	*Ounces per ton*	*Ounces per ton*
Au	0.075							0.03
Ag	35.35		8.2		37.0	6.0	5.4	10.0

A, from western ores [a]; B, from Joplin ores [a]; C, source of ore not known [a]; D, from Joplin ores [a]. E. from Butte ores [b]; F, from Cœur d'Alene ores [b]; G and H, source of ore not given.[c]

[a] Personal communication to the author.

[b] Stock, K, Sintering zinc residues: Chem. and Met. Eng., vol. 20, May 15, 1919, pp. 525–528.

[c] Johnson, Edward Mackay, Recovery of spelter and the treatment of retort residues: Met. and Chem. Eng., vol. 18, Feb. 1, 1918, pp. 135–139.

METHODS AVAILABLE FOR THE TREATMENT OF RETORT RESIDUES

The methods that have been used or suggested for the treatment of retort residues may be classified as follows:

- **I. Smelting directly in the lead blast furnace for the recovery of lead, copper, and precious metals.**
  - ***a.* Residues mixed in small proportions with ore charge.**
  - ***b.* Residues smelted alone with the necessary fluxes.**
- **II. Burning and sintering to remove carbon and furnish a desirable product for the lead blast furnace, followed by smelting in a blast furnace.**
- **III. Dry screening.**
  - ***a.* For the recovery of coke.**
  - ***b.* For the concentration of lead and precious metals into an enriched product.**

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[3] Schuchard, E., Beitrag zur Metallurgie des Zinkes: Metallurgie, Jahrg. 2, Mar. 8, 1905, pp. 105–109.

[4] Ingalls, W. R., The metallurgy of zinc and cadmium. 2d ed., 1906, p. 534.

IV. Magnetic separation.
V. Wet concentration.
  a. For the recovery of lead and precious metals only.
  b. For the recovery of zinc and coke.
VI. Burning on Wetherill grates for the production of zinc oxide.
VII. Miscellaneous processes.

DIRECT SMELTING IN THE LEAD BLAST FURNACE FOR THE RECOVERY OF LEAD, COPPER, AND PRECIOUS METALS

When zinc ores containing lead, copper, gold, and silver are subjected to distillation most of these metals remain in the residues. According to Ingalls,[5] in a test on Butte and Superior ore, extending over a period of about two years, the weight of the residue was 46.75 per cent of the weight of the raw ore. The residue contained 88.14 per cent of the gold, 77.4 per cent of the silver, 78.5 per cent of the copper, and 47.75 per cent of the lead in the raw ore. This ore assayed 50.5 to 53.2 per cent Zn, 2.4 to 4.7 per cent Fe, 0.22 to 1.0 per cent Mn, 2.4 to 3.6 per cent Pb, 0.33 to 0.50 per cent Cu, 11.6 to 14.1 per cent insoluble, and 0.03 ounce Au and 14.0 to 18.5 ounces Ag per short ton. Two and one-half tons of ore, containing 40 ounces of silver, yielded about 1 ton of zinc. About 35 per cent of this zinc was derived from the first draw, 50 per cent from the second draw, and 15 per cent from the third draw. Zinc from the first draw assayed 0.164 ounces Ag per ton and 1 per cent Pb; from the second, 0.583 ounces Ag and 1.5 per cent Pb; from the third, 1.178 ounces Ag and 2.25 per cent Pb.

When the residue contains enough copper, lead, silver, and gold to cover the treatment charge and freight and leave a profit, the usual practice is to ship it just as it is to a lead smelter. Because of its fineness and its zinc content this residue is not a desirable feed for a lead blast furnace and it is usually charged in small amounts with more easily smelted material.

E. M. Johnson has described, however, a blast furnace run made in 1903 at the plant of the Cherokee Lanyon Smelter Co., Gas, Kans., in which residues made up the chief burden of the furnace.[6] The furnace measured 36 by 90 inches at the crucible and 14 feet from center of the tuyères to the feed floor. Because of the small size of the furnace and the high percentage of zinc in the charge, the furnace had to be "let out" frequently so that the accretions could be barred down. The longest continuous run was six weeks. An analysis of the residue used is given under H, Table 1. Enough iron ore, limestone, siliceous ores, and galena concentrates were charged with the residue to give a suitable slag and a 9 per cent lead fall, to collect the gold and silver. During operation for seven months 45.6 tons of charge were smelted per day, with a saving of 90.8 per cent of the silver, 92.4 per cent of the gold, and 92 per cent of the lead.

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[5] Ingalls, W. R., Metallurgy of zinc: Mineral Industry, vol. 27, 1918, p. 773.

[6] Johnson, E. M., Smelting-retort residues made from western zinc ores: West. Chem. Met., vol. 1, August, 1905, pp. 179–181; Recovery of spelter and the treatment of retort residues: Met. Chem. Eng., vol. 18, Feb. 1, 1918, pp. 135–139.

Table 2.—*Summarized results of blast furnace run on retort residues—results actually obtained based upon 149 furnace days*

	Per day	Per furnace day	
Labor and overhead expense	$71.00	$117.20	
Sampling and assaying	2.30	4.18	
Supplies	2.00	3.72	
Repairs and renewals	4.90	9.00	
	80.20		$134.10
	Tons per day	Minus margins	
Galena	2.8	× −6.05 = −16.94	
Lime rock	6.5	× −1.09 = −7.09	
Iron ore	6.7	× −4.25 = −28.48	
Coke	6.1	× −8.20 = −50.02	
	−22.1		−102.52
			−236.63
	Tons per day	Plus margins	
Residue	13.3	× +9.34 = +124.22	
Ores	12.2	× +3.76 = + 44.87	
	25.5		+169.09
Loss per day			−67.54
Loss per month			2,026.20
Loss per ton of residue			− 5.07

The experiment extended over a total period of 273 days, during which time there were only 149 actual furnace days. The financial result of the experiment was a loss of $5.07 per ton of residue smelted. The itemized costs and margins appear in Table 2. Johnson estimated that under more favorable conditions this loss could be changed into a profit, but he thought the saving over shipping the residues to a lead smelter would be doubtful.

BURNING AND SINTERING TO REMOVE CARBON AND GIVE A DESIRABLE PRODUCT FOR THE LEAD BLAST FURNACE

An appreciable saving in freight charges can be effected by burning out the carbon in the residues before shipping them to a lead smelter. This process incidentally eliminates some of the zinc, concentrates the lead, copper, gold, and silver, and yields a sintered product much more satisfactory as an addition to the blast-furnace charge than the unburned residues.

Leslie[7] and Stock[8] have described the sintering method used at the zinc smelters of the American Metal Co. at Bartlesville, Okla.

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[7] Leslie, E. H., Zinc smelting at Bartlesville, Okla.: Min. Sci. Press, vol. 109, July 11, 1914. pp. 44–49.

[8] Stock, K., Sintering zinc residues: Chem. and Met. Eng., vol. 20, May 15, 1919, pp. 525–528.

The residues were selected so that the resulting sinter would have a lead content as much over 5 per cent as possible, because 5 per cent was the minimum limit of payment according to smelter schedules. A typical charge consisted of the Butte and Coeur d'Alene residues, the analyses of which are given in columns E and F, Table 1, mixed in the proportions in which they came from the retorts. When used alone the Coeur d'Alene residue softened and did not sinter well because of its higher lead content. The residue was burned in clinkering beds 47 to 50 feet square at the bottom, and arranged in a long row which was skirted on one side by an air main and on the other by tracks for locomotive crane and railroad cars.

In starting a bed the site was first carefully leveled and a series of air channels was built of loose brick, covering the area like a gridiron, as shown in Figure 1.[9] At the rear center of the pile a long-radius bend of 12-inch pipe led from the air main to a breeching built of brick or tile bats, which were cemented and then covered with earth to prevent air leakage or damage during the removal of clinker. This breeching divided the air current into a header leading to right and to left along the back of the pile, parallel to the air main. The central third of this header was formed by two walls of closely dry-laid brick, four courses high and 4 inches apart, roofed with brick or tile laid closely together. The end portions of the header were similar but only three courses high. At intervals of 3½ feet a half brick was removed from the front wall, and from these openings—13 in number—the minor channels led completely across the pile toward the front. These channels were also built of dry brickwork, but the bottom courses consisted of half bricks. This arrangement provided a large number of openings through which the air blast could find its way into the pile above. All these channels were covered with a thin ridge of earth patted down with a flat shovel, and the area was then ready for its charge of residue.

The retort residues were brought to the heaps either in tramcars on an elevated track that traversed the site of several piles or in standard gondolas. Cars filled with the residues were switched to the site of a new pile, and about 100 tons were unloaded by a locomotive crane into a ridge at the rear of the bed covering the back header, with the toe of the slope extending a few feet beyond the channels, as sketched in Figure 1. Two more carloads were piled along the front of the heap, leaving a trough running across the middle of the bed, where all of the 13 parallel channels remained uncovered. Two more carloads were then spotted ready for the heap. An armful of coarse wood was deposited in the valleys remaining between the exposed portions of the air channels; a shovelful of hot coals was placed on the wood, and the whole was covered with residue from the adjoining slope.

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[9] Stock, K., Sintering zinc residues: Chem. and Met. Eng., vol. 20, May 15, 1919, p. 526.

When a row of fires had been kindled in this manner, a little air was admitted by raising a slide gate in the pipe leading to the header and the two cars of residue were unloaded into the trough, making a square pile about 5 feet deep. While this was being done the air gate was gradually opened wide, and the combustion spread through the mass in all directions. Within a week the fire had extended to the entire area of the pile and the center had settled somewhat. The pile was kept leveled and any blowholes were smothered under piles of fresh residues. As the combustion proceeded and the pile settled, four to six extra cars of residues were spread over the surface, making a total of 500 to 600 tons in one heap. The burning of a pile required about 35 days. When burning was completed the air was

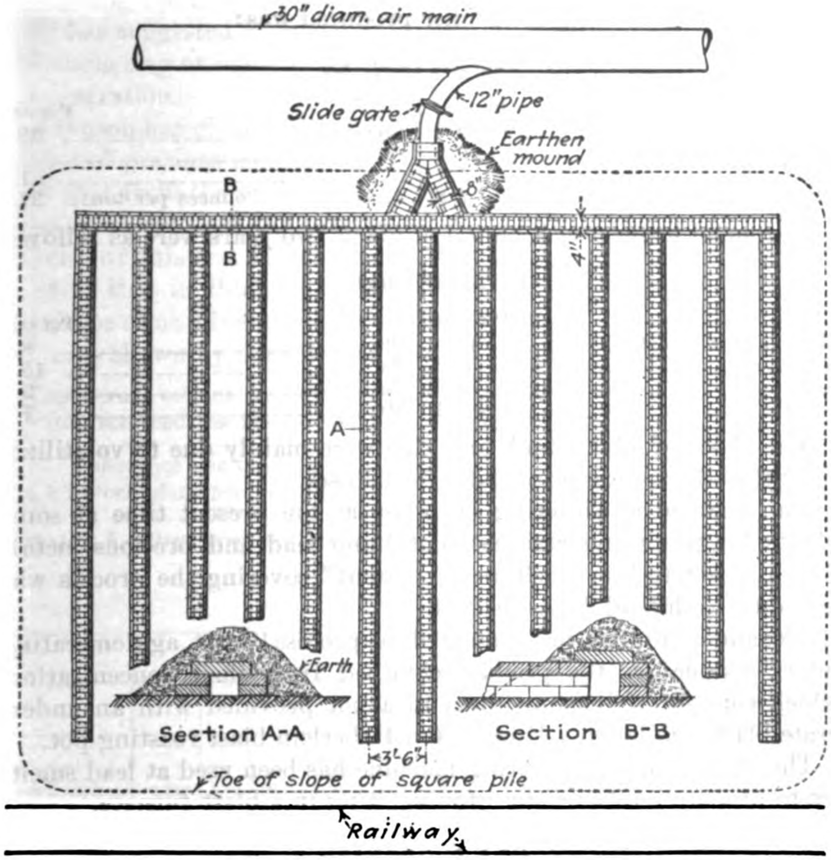

FIGURE 1.—Plan of clinkering heaps

shut off and the pile was wet for two or three days with a garden hose. When it was cool enough the clinkered mass was broken up by blasting, dug out with sledge, pick, and shovel, and piled to one side on clean ground whence it was loaded into cars by a locomotive crane with a clamshell bucket.

The sintering plant at Bartlesville consisted of 17 such "kilns," and prepared about 2,100 tons of clinker per month. The cost of sintering was $1.53 per ton of clinker produced.

A typical analysis of the clinker produced was:

Analysis of clinker

	Per cent		Per cent
Zn	12.0	SiO_2	38.0
Fe	12.9	Cu	.8
Mn	1.1	C	1.2
Pb	9.8	Ag ounces per ton	27.0

The metal losses over a period of about two years were as follows:

Metal losses

	Per cent		Per cent
Shrinkage in weight	32.0	Pb	8.9
Au	2.8	Zn	18.1
Ag	3.2	Fe	2.8
Cu	+1.4	Mn	7.2

The losses of silver, lead, and zinc were mainly due to volatilization; the others were principally dust losses.

This method of sintering is in use at the present time at some plants for preparing residues containing lead and precious metals for shipment to lead smelters. A patent[10] covering the process was issued to Archibald Jones in 1914.

Alexander Folliet has patented a process[11] for agglomerating retort residues or the product obtained from their concentration, which consists of burning them in a pot provided with an undergrate blast, similar to a Huntington-Heberlein blast roasting pot.

The Dwight-Lloyd sintering machine has been used at lead smelters to prepare retort residues for smelting in a blast furnace.

DRY SCREENING

It is often possible to remove a large part of the unburned coal or coke from retort residues by merely screening on a coarse screen, since most of the particles of coke are larger than the particles of ore residue. By this removal of the coke the zinc, lead, silver, and gold assays of the remainder of the residue are at the same time raised. The object of screening may be mainly to furnish a product containing sufficient lead and precious metals to make profitable its shipment to a lead smelter; or residues from ores such as those from the Joplin district, which do not contain lead and silver, may be screened merely to recover their fuel content.

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[10] Jones, Archibald, Treatment of zinc residues, United States patent 1,112,010, Sept. 29, 1914.

[11] Folliet, Alexander, Agglomerating process. United States patent 1,029,182, June 11, 1912; Procédé de traitement agglomérant par réaction des sous-produits d'usines, ferrugineux et plombifères, obténus sous forme de cendrées, en vue de leur traitement final pour réduction au four à cuve, French patent 428,570, June 24, 1911.

A satisfactory separation can not be made by screening if the fuel and ore residue have been sintered together during the process of extracting the zinc or by subsequent oxidation or weathering. Gerlach has suggested [12] crushing the residue slightly to free the fuel of adhering slag or oxide films before subjecting it to screening or other concentration.

Johnson has given the results of a screen test on residue containing lead and precious metals,[13] in which the residue from the ore charge and that from the blue-powder charge were screened separately on a 3-mesh screen, and the products were weighed and analyzed. The results of this test are shown in Table 3. From these results it is evident that in this case to screen the residue from the ore charge would be of no advantage, but to screen the blue-powder residue would be, as is shown by the following calculations:

2.3 tons crude residue per day per block, at $0.61	$1.40
1.5 tons screened residue per day per block, at $2.46	3.69
Difference per day	2.29
On a 5-block plant per month	345.00

Table 3.—*Dry screening test of retort residues, 3-mesh screen, 7/32 inch opening*

Mesh	Pounds	Tons	Per cent	SiO_2	Fe	Zn	S	Pb	Ag	Au	Gross value	Treatment, freight, etc.	Net value
Charge residue													
				Per cent	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Oz. per T.*	*Oz. per T.*			
Plus 3	3,819	1.91	32.3	13.3	10.0	6.0	3.8	2.5	7.8	0.16	$7.04	$5.70	$1.34
Minus 3	8,025	4.01	67.7	10.6	13.4	10.0	5.5	7.21	1.7	.27	13.01	5.70	8.21
		5.92	Crude residue=72 per cent								11.69	5.70	5.99
Blue-powder residue													
Plus 3	1,617	0.81	35.2	18.0	7.4	6.2	2.1	2.2	3.7	0.09	$2.80	$5.70	—$2.90
Minus 3	3,008	1.50	64.8	11.1	9.4	8.7	3.8	6.2	6.7	.14	8.16	5.70	2.46
	4,625	2.3	Crude blue-powder residue=28 per cent								6.31	5.70	.61

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[12] Gerlach, Oscar, Process for economizing metallic reductions, United States patent 1,313,287, Aug. 19, 1919.

[13] Johnson, E. M., Recovery of spelter and the treatment of retort residues: Met. and Chem. Eng., vol. 18, Feb. 1, 1918, pp. 135–139.

In 1913 and 1914, according to Ingalls,[14] the Edgar Zinc Co. at Cherrydale, Kans., were recovering the fuel from residues by screening and, after mixing it with 20 per cent fresh coal, burning it under steam boilers. The residues were first sifted over a screen with five-eights-inch openings. The oversize, being high in zinc, was returned to the distillation furnaces; the undersize was sifted over a 20-mesh screen. The oversize from the 20-mesh screen was burned under boilers by using perforated grates and undergrate blast. Some zinc oxide was recovered from the rear of the combustion chamber and was saved.

Several plants are at present screening the residues from Joplin ores for the recovery of the unburned fuel, which is then used as reduction fuel in the new charge for the retorts, or as fuel for the Wetherill grates in a zinc oxide plant. As an example of this practice, at one plant the residue is first dumped on a grizzly with 1 to 1½ inch openings. This grizzly takes out the large chunks of slaggy material which are discarded. About 90 per cent of the residue goes through the grizzly; this undersize is then screened on a 6-mesh screen. About 25 per cent of the undersize from the grizzly goes through the 6-mesh screen. This minus 6-mesh product, containing about 25 per cent zinc, is not being further treated at present, but is being stored for possible utilization. The oversize from the 6-mesh screen is sifted over a screen with one-half-inch openings; the oversize from the latter is discarded and the undersize, amounting to about 65 per cent of the total weight of the residue, is reused as reduction fuel in the retort charge.

The continual reuse of the recovered fuel gradually reduces the carbon content of the retort charge until it may fall as low as 20 per cent. When this point is reached the residue is discarded and a fresh start is made with new coal as reduction fuel. With this method of working, the weight of ore charged per retort has been increased by 30 to 50 per cent. The residues run higher in zinc than formerly, but, because of the smaller weight of residues discarded, less zinc is lost. The residues are more slaggy than when fresh reduction fuel alone is used, but since the ores smelted are low in iron the slag does not have a bad effect on the retorts, and the extra work of cleaning the retorts is counterbalanced by the greater amount of ore that can be charged.

At another plant, at which operating conditions are similar to those existing at the plant just mentioned, the better parts of the residue are selected and reused as charge fuel without preliminary screening. Formerly 50 per cent of clean reducing fuel was mixed with the ore charge, but the present practice is to use 30 per cent of reduction fuel made up of six-sixteenths selected retort residue and ten-sixteenths dead coal. The residues, instead of being blown out of the retorts, are shoveled and scraped out. As in the preceding example, the residues assay higher in zinc than under the old practice, but the net zinc loss is less because of the smaller weight of residues discarded; and the heavier charge made possible by the new practice makes up for the extra work of cleaning the retorts.

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[14] Ingalls, W. R., Metallurgy of zinc in 1913: Min. Ind., vol. 22, **1914**, pp. 805–807; Metallurgy of zinc in 1914; Min. Ind., vol. 23, 1915, pp. 794–798.

At a third plant the residues were formerly screened on a 6-mesh screen and the oversize, which contained about 70 per cent of carbon but very little zinc, was reused as reduction fuel, though only to the extent of 15 per cent of the total reduction fuel used. Higher proportions than this were unsatisfactory because of the slaggy retorts produced and resultant labor troubles with the furnace men. At present this screening plant is not running, but the fuel recovered while it was in operation is still being used.

A similar screening plant at another smelter separates 60 per cent of the residue as a product containing 75 per cent carbon. This is used as fuel for Wetherill grates by the zinc oxide department, which pays for it according to its carbon content.

Some other companies that have tried screening their residues have not been satisfied with the results; they believe that poorer zinc recoveries resulted from the reuse of the recovered fuel and that this zinc loss and the extra labor entailed were not counterbalanced by the value of the fuel saved.

MAGNETIC SEPARATION

In 1913 Lindt reported[15] the results of some experiments in the electromagnetic concentration of retort residues containing 3.8 per cent Zn, 0.56 per cent Pb, 18.9 per cent Fe, 4.73 per cent S, and 1.14 ounces Ag per ton. From this material he separated 26 per cent as a magnetic concentrate containing 3.8 per cent Zn, 0.16 per cent Pb, 59.6 per cent Fe, 3.65 per cent S, and 3.38 ounces Ag per ton, leaving 74 per cent of nonmagnetic material containing 3.8 per cent Zn, 0.71 per cent Pb, 4.6 per cent Fe, 5.12 per cent S, 0.46 ounce Ag per ton.

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[15] Lindt, —, Zur Aufbereitung der Zinkmuffrückstände: Met. und Erz, vol. 10, Mar. 22, 1913, pp. 347–348.

TABLE 4.—*Compilation of magnetic separation tests of retort residue—screen-sized feed*

	Weight	Per cent of feed	Per cent of original	Analysis					Recovery from original feed, per cent				
				Ag	Pb	Zn	Insoluble	C	Ag	Pb	Zn	Insoluble	C
Original Feed													
	Gms.			*Oz.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*					
Original feed	9,700.0	------	100.0	7.4	4.1	6.1	20.7	32.0	100.0	100.0	100.0	100.0	100.0
Concentrates													
Minus 4+20-mesh first concentrate	1,907.3	28.0	19.6	17.40	8.27	5.66	25.60	3.20	45.6	40.0	18.3	24.3	2.0
Minus 20+40-mesh first concentrate	702.8	42.6	7.2	17.80	9.75	7.46	22.80	.40	17.4	17.3	8.9	8.3	.09
Minus 40-mesh first concentrate	427.3	34.2	4.4	16.88	7.88	10.60	20.40	.20	10.0	8.5	7.7	4.3	.03
Total	3,037.4	------	31.2	17.3	8.6	6.8	24.4	2.1	73.0	65.8	34.9	36.9	2.12
Middlings													
Minus 4+20-mesh second concentrate	671.5	9.8	6.9	12.4	7.0	4.97	36.20	9.80	11.6	11.9	5.7	12.1	2.1
Minus 40-mesh second concentrate	148.1	11.8	1.5	7.2	7.2	11.30	26.8	7.10	1.5	2.7	2.8	2.0	.3
Total	819.6	------	8.4	11.5	7.04	6.1	34.5	9.3	13.1	14.6	8.5	14.1	2.4
Tailings													
Minus 4+20-mesh tailings	4,212.7	62.1	43.4	1.50	0.88	4.41	15.2	55.0	8.8	9.4	31.5	32.0	75.5
Minus 20+40-mesh tailings	936.6	56.7	9.6	1.75	4.35	7.02	20.00	49.20	2.3	1.0	11.1	9.3	14.8
Minus 40-mesh tailings	607.6	48.6	6.3	3.00	5.34	11.78	24.00	25.00	2.5	8.2	12.1	7.3	4.9
Total	5,756.9	------	59.3	1.71	1.28	5.62	16.80	51.3	13.6	18.6	54.7	48.6	95.2

A magnetic separation test of the residues was made by J. T. Norton, of the Intermountain experiment station of the Bureau of Mines; the analysis is given under column C, Table I. A sample of 10,000 grams previously crushed to pass a 4-mesh screen was sized on 20 and 40-mesh screens; the minus 4 plus 20, the minus 20 plus 40, and the minus 40 products were fed separately to a Campbell magnetic separator, which had an inclined vibrating conveyor and take-off belts under magnets for the discharge of concentrates. It had three magnets of different strengths. The speed of the take-off belts was about 100 feet per minute. Table 4 summarizes the results of this test.

The combined concentrates and middlings weighed 3,857 grams, representing 38.57 per cent of the original weight, and assayed 16.1 ounces Ag per ton, 8.2 per cent Pb, 6.6 per cent Zn, 26.6 per cent insoluble, and 3.6 per cent carbon. They contained 86.1 per cent of the silver, 80.4 per cent of the lead, 43.4 per cent of the zinc, 51 per cent of the insoluble, and 4.5 per cent of the carbon in the original sample of residue treated. Over 50 per cent of the zinc and 95 per cent of the carbon were eliminated in the tailing.

In 1919 a patent[16] was issued to Oscar Spitzer and C. C. Conover on a method of treating zinc-retort residues by magnetic separation in which especial emphasis was placed upon the desirability of cooling the residues rapidly—either naturally or in an artificial cooling apparatus—and feeding them to the magnetic separator before the contained metals had oxidized and lost their magnetic permeability. With the specific apparatus described the residues were first cooled in a reducing atmosphere in a cylinder which was closed except for the feed and discharge pipes and which revolved partly submerged in a tank of water. The cooled residues passed through a magnetic separator. The tailing from this separator was sized on a 12-mesh screen to take out the coarser particles of carbon and the undersize was sent to a second more powerful magnetic separator where a second concentrate was made. The tailing from the second separator was sized on a 20 to 24 mesh screen to remove the smaller particles of carbon.

A modification of this process was used for some time for residues from western ores and gave satisfactory results. In practice the residues were run through a coarse trommel to remove chunks of slag, and the undersize from the trommel was dried and fed to a magnetic separator. This made a magnetic concentrate containing the metals of value (lead, gold, and silver). The tailings from the magnetic separator were screened. The carbon remained on the screen and was reused as charge coal; the undersize was discarded.

WET CONCENTRATION

FOR RECOVERY OF LEAD AND PRECIOUS METALS

The enrichment, by jigging, of retort residues containing lead and silver has long been practiced. Residues from lead-free ores can sometimes be profitably concentrated for their zinc and coal contents, but until recent years most attempts to concentrate such residues were not successful. Some of these attempts were no doubt unsuccessful because of improperly designed concentrating mills; but in many residues the zinc is so intimately associated with particles of slag or ash that the making of a high-grade zinc concentrate is impossible.

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[16] Spitzer, Oscar, and Conover, C. C., Treatment of zinc retort residues, United States patent 1,315,349, Sept. 9, 1919.

TABLE 5.—*Typical analyses of concentrates from lead-bearing retort residues*

	A residue	A concentrate		B residue	B concentrate	G residue		G concentrate	H residue	H concentrate
		First class	Second class			(a)*	(b)*			
	Per cent	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*
Zn	4.00	3.87	5.00	2.50	6 to 7	2.60	3.94	6.89 to 7.70	4.00	8.00 to 12.0
Pb	5.00	24.25	13.16	8.50	40 to 45	2.13	11.60	35.00 to 57.00	10.00	30.0 to 35.0
Fe	16.55	42.75	20.68	14.50	15 to 20	15.72	11.19	11.26 to 21.22	?	
Ag	.016	.032	.016	?			.035	0.04 to 0.07	.01	0.04 to 0.05
Cu	.05			?			.19			
Cd	Nil.			?						
As	Nil.			?						
Sb	Nil.			?			.25			
S	?			4.00		.62	1.72		?	
CaO	2.50			2.50		4.00	1.33		8.00	
MgO	.45			1.50		2.16	.73		?	
SiO_2	50.00	18.66	44.67	60.00	15 to 20	34.30	32.23	13.56 to 15.38	17.50	
C						30.02	30.40		?	
Al_2O_3						2.60	1.10		?	

* Residuum from different mixtures of ore.

Some typical analyses of lead-bearing residues and of the concentrates produced from them, as given by Firket,[17] are shown in Table 5.

Johnson has reported[18] the results of some experiments in concentrating lead and silver bearing residues by means of a Hancock jig. The cost of concentrating, based on feeding 100 tons of crude residue per 24 hours, was 24 cents per ton for new residue, or 35 cents, including loading and hauling, for old residue. These were pre-war costs, and would, of course, be greater under present conditions. The old, or weathered, residue was more difficult to concentrate than residue direct from the retorts. Table 6 gives the results of one of the test runs reported by Johnson.

TABLE 6.—*Result of jigging test of lead and silver bearing residues*

	Analyses									Screens	
	SiO_2	Fe	Mn	Zn	S	Pb	Ag	Fixed carbon	Volatile matter	Mesh	Wire
	Per cent	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Per cent*	*Oz. per ton*	*Per cent*	*Per cent*		
Crude residue	11.0	29.7	0.84	5.5	3.1	3.4	5.4				
Minus 5-mesh	7.6	36.7	.94	6.7	3.7	4.0	8.0				
Hutch No. 1	5.6	58.9	1.06	7.0	4.0	5.8	11.0			6	14
Hutch No. 2	7.8	52.1	1.20	9.2	5.2	4.3	11.0			7	16
Hutch No. 3	18.6	31.1	1.00	7.7	4.0	2.8	9.3			6	14
Hutch No. 4	15.6	12.5		4.0		1.5	4.6			6	14
Tailings	13.2	7.4		3.0		.7	3.6				
Return to rolls	25.0	4.8		3.4			2.3			Ash.	
Coal material		2.7			.7			70.8	2.0	27.2	
Recovery, per cent		63.5				52.7	74.5				

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[17] Firket, Ad. Annales des Mines de Belgique. t. 6, 1901, p. 1; Ingalls, W. R., The metallurgy of zinc and cadmium. 2d ed., 1906, pp. 537 and 539.

[18] Johnson, E. M., Recovery of spelter and the treatment of retort residues: Met. Chem. Eng., vol. 18, Feb. 1, 1918, pp. 135–9.

Weights and values

	Dry weight, pounds	Value, per ton	Cost, per ton	Net, per ton	Total realization
Crude residue	61, 109	$5. 11	$5. 70	−$0. 59	−$18. 00
Minus 5-mesh	46, 076	7. 58	5. 80	1. 78	40. 94
Hutch No. 1	11, 250				
Hutch No. 2	7, 260				
Hutch No. 3	4, 640				
Hutch total	23, 150	11. 08	6. 75	4. 33	50. 01
Hutch No. 4	8, 050				
Tailings	21, 650				
Slimes, etc	8, 259				

The test shows that although this residue in its crude state could not be shipped profitably to a lead smelter, a concentrate worth shipping could be made from it, either by screening on a 5-mesh screen or by jigging, with a difference of about 30 cents per ton in favor of the latter. Johnson concluded from his tests that when the crude charge residue showed an appreciable net value jigging it did not pay; the blue powder residue, however, concentrated better and concentration might be profitable whether or not the residue showed a net value in the crude state.

At the Salt Lake Station of the United States Bureau of Mines tests of the same residues that were used for the magnetic separation tests previously described indicated that a high recovery could not be obtained from these residues by wet methods—whether simple classification, jigging, tabling, or oil flotation—though a somewhat enriched concentrate could be made with a recovery of 50 to 60 per cent of the lead and silver.

FOR RECOVERY OF ZINC AND COKE

Steger,[19] in washing two residues containing respectively 5.16 per cent and 5.36 per cent of zinc, obtained concentrates assaying 13.80 per cent and 14.60 per cent of zinc respectively, of which 7.92 per cent and 8.15 per cent respectively, were in the form of ZnO. Ashes amounting to 7,063 kg. were washed for unburned coal, yielding 1,505 kg. of 4 to 30 mm. size, but at a cost that would have made the operation unprofitable.

G. A. Chapman has proposed to separate the carbon from retort residues by oil flotation.[20] He cites as an example that from a residue containing 30.1 per cent carbon and 10 per cent lead he obtained a float concentrate assaying 72.5 per cent carbon and 0.75

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[19] Steger, —, Aufbereitung von Zinkhüttengekrätz: Ztschr. Berg-Hüt. Salinen., Bd. 44, 1896, p. 147; Ingalls, W. R., The metallurgy of zinc and cadmium, 2d ed., 1906, p. 540.

[20] Chapman, G. A., Improvements in the separation of certain nonmetallic from other mineral substances, British patent 8,567, Apr. 9, 1908.

per cent lead, containing 89 per cent of the carbon originally present in the residue. Most of the lead remained in the part of the residue that did not float.

Stolzenwald proposed to size residues in trommels and separate the zinc particles by blowing.[21]

At present a few companies are successfully concentrating the residues from lead-free ores. One company has found that the method is applicable to the residues from particular ores only and that residues from Joplin ores can not be satisfactorily concentrated. Others have obtained good results with Joplin ores, making a concentrate high enough in zinc to return to the retorts and recovering the unconsumed carbon as a clean product suitable for reuse as charge coke. Another company formerly screened its residues on a 6-mesh screen, obtaining an oversize, containing about 70 per cent carbon, that was reused as charge coal and a minus 6-mesh product that was concentrated on tables. A concentrate assaying about 35 per cent zinc was obtained; it was so difficult to smelt that the concentrator proved unprofitable and was shut down.

The inconsistent results obtained in the various attempts to concentrate residues—both from lead and silver bearing and from lead-free ores—are perhaps due to differences in the residues caused by differences in the ores smelted or the reducing fuel used, or to differences in smelting practice; or they may be partly due to differences in equipment or operation of the concentrators.

At one plant where the concentration of residues from Joplin ores has been worked out successfully the concentrating equipment consists of Joplin jigs, James tables, and the necessary screens, elevators, and auxiliaries to make a well-balanced and efficient mill. The residues are crushed and sized; the finer sizes are sent to the tables and the coarser sizes to the jigs. A concentrate carrying 48 to 72 per cent zinc is obtained, which is used as " stuffing " to close up the ends of the retort condensers. After the charge is distilled this " stuffing " is scraped out of the condensers and mixed with the blue powder. It is then charged to the blue-powder retorts with the blue powder and the concentrates from broken condensers. See page 21. In this way the concentrates are smelted without difficulty. The concentrates can also be roasted, mixed with reduction fuel, and smelted in the same way as roasted ore.

The coke in the residue floats on the overflow water from the jigs and is skimmed off as a product containing 12 to 15 per cent ash. It is screened to eliminate the fines, and the oversize is used as charge coke. The tailings from the mill assay 2 to 3 per cent zinc.

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[21] Stolzenwald, Gustav, Verfahren zur Aufbereitung von Zinkhüttenrückständen, German patent 198,577, May 20, 1908.

Both new residues and those accumulated from past operations are concentrated with good results.

At another plant residues are concentrated in a similar manner, except that the management has decided that the extra recovery obtained by tables does not pay for their operation, and now uses jigs alone. The dry residues from current operations are screened to separate the coke before they are fed to the mill.

CAUSE OF FAILURES

Some of the failures to treat residues successfully by jigging and tabling have possibly been due to an unwillingness to risk the capital outlay required to build an efficient mill. Those who are now successfully concentrating residues have built complete concentrators at considerable expense and are convinced that too much saving in the first cost of a concentrating mill will preclude satisfactory results.[22]

BRIQUETTING OF RESIDUE CONCENTRATES FOR SMELTING

Tests were made in 1921 at the Mississippi Valley station of the Bureau of Mines in an endeavor to overcome some of the difficulties met in smelting the zinc concentrates obtained by jig and table concentration of retort residues.

The difficulties encountered in smelting these concentrates are rapid deterioration of retorts and low recovery of metal. It was thought possible that by mixing the concentrates with pitch and coke and making them into briquets, such as those used in the Fulton electrothermic distillation furnace for zinc ores,[23] the corrosion of retorts could be prevented and the recovery of zinc improved.

The object of briquetting was to hold the material in a matrix of coke that would keep its shape during and after distillation, thus preventing the formation of slag and holding any FeS formed so that it could not attack the retort. If enough iron oxide were in the charge the iron would decompose the ZnS present; this fact and the intimate mixture of the carbon with the ore would be conducive to high extraction of zinc. A typical analysis of the residue concentrates used in the test was: Zn, 36 per cent; Pb, 2.1 per cent; CaO, 3.1 per cent; Fe, 9.5 per cent; S, 3.2 per cent.

As a first step, laboratory tests were carried out as follows: The concentrates were mixed with 40 to 60 per cent of their weight of coke and 15 to 20 per cent of coal-tar pitch. This mixture was warmed till plastic; it was then pressed in warm molds to form briquets 1 inch in diameter and 1½ inches long. These briquets

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[22] Since this survey was made several new residue concentrating mills similar to the ones above described have been built at Oklahoma zinc smelters.

[23] Fulton, C. H., Electric-resistance furnace of large capacity for zinc ores: Trans. Am. Inst. Min. and Met. Eng., vol. 64, 1921, pp. 188–226.

Original from UNIVERSITY OF ILLINOIS AT URBANA-CHAMPAIGN

were surrounded with crushed coke and baked at 600° C. to drive off volatile hydrocarbons; then they were heated to 1,300° C. to distill the zinc. Some of the briquets were distilled without the preliminary baking. The extraction in these small-scale tests ranged from 96 to 99 per cent, as indicated by analyses before and after distillation.

One ton of the residue concentrate was then made into 40-pound briquets 7½ inches in diameter and 16 to 17 inches long and shipped to the smelter for distillation. The coke and pitch used in these briquets were crushed to pass a ¼-inch mesh screen. Ninety per cent of the concentrates would pass a ¼-inch mesh screen. In seven out of nine tests with these large briquets the extraction was 90.7 to 97.5 per cent, but because of certain adverse conditions the zinc recovery was only 50 to 70 per cent. The tests were made in a newly fired furnace; the front ends of the retorts were cold, and in consequence much of the zinc condensed in the front of the retort instead of in the condenser. Another reason for loss was that the condensers were not put in place until the volatile matter from the pitch was driven off; consequently much zinc was burned. It should be possible to bake the briquets with the condensers in place; then, with proper temperature conditions in the retorts, good recoveries of zinc should be possible. Since these tests were made, however, it has been found possible to smelt the residue concentrates by simpler methods; consequently, the experiments with the briquets have never been carried to a conclusion.

BURNING ON WETHERILL GRATES FOR THE PRODUCTION OF ZINC OXIDE

At smelters where both retort furnaces and zinc-oxide plants are operated it is common practice to burn the retort residues, either alone or mixed with the regular ore charge, on Wetherill grates for the production of zinc oxide. If the residues are burned alone the unconsumed coal they contain suffices, or nearly suffices, to supply the requisite heat and very little addition of other fuel is necessary.

Operators of some plants have found that the oxide obtained by burning zinc residues is unsuitable for the paint or rubber trade, because of a high content of zinc sulphate, irregular composition, and poor color. At other plants the quality is perhaps slightly poorer than that of oxide produced from ores, but not enough to prevent the two oxides being mixed without detriment to the price of the better-grade oxide. At times the oxide from residues is as good as or better than the oxide from ores. These differences in quality of the oxide produced from residues at different smelters are perhaps partly due to differences in the ores treated in the retorts.

Naturally, when high-grade lead-free oxide is being made only residues from lead-free ores can be used in the oxide plant. When residues containing lead and silver are burned on Wetherill grates most of the lead is volatilized and forms lead sulphate or basic lead sulphate, thus giving a leaded zinc oxide. If sodium chloride is present in the residues nearly all the silver will be volatilized; in the absence of sodium chloride 75 to 80 per cent of the silver will remain in the clinker.[24]

When residues are burned alone the extraction of zinc ranges from 50 to 85 per cent. When the residues are added to the regular ore charge the Wetherill grate clinker usually retains no more zinc than when they are not added.[24a]

The profit in burning retort residues depends, of course, on the zinc content of the residues and the price of zinc oxide; sometimes it is more profitable to screen the coal from the residues and use it as fuel in the ore charge to the Wetherill grates, even though in this way most of the zinc is discarded.

Hofman describes the practice in burning residues alone. When the residues contain 6 to 8 per cent zinc, a grate 6 by 12 feet will receive a charge of 3,500 to 3,700 pounds, which makes a bed 12 inches deep; with 10 per cent zinc the charge will be about 2,700 pounds. Crowding the grate in this way reduces the yield of zinc to about 50 per cent, though the lead is completely volatilized. The sulphur content of the residue should be under 2 per cent if the lead content is low, or the oxide produced will be high in $ZnSO_4$ and SO_2. If the residue contains considerable lead, as much as 3 or 4 per cent sulphur can be present, for it combines with the lead as $PbSO_4$.

A heavier blast and a higher temperature are required for burning residues than for burning ores. A 12-grate block with 6 hours to a charge will treat 60 tons residue in 24 hours. In good work the clinker contains 1 to 5 per cent zinc and only a trace of lead.

Hofman states that the residues from Missouri and Wisconsin ores give an oxide with 6 to 15 per cent $PbSO_4$; from Butte ores, an oxide with 20 to 35 per cent $PbSO_4$, often 2 to 6 per cent $ZnSO_4$ and 0.2 to 1 per cent SO_2. Impure oxide may be added to the charge to the spelter furnace or sold to manufacturers of lithopone.

Since the first large proportion of the zinc in a retort charge is so much more easily distilled than the last few per cent, it has been suggested that the retorts be charged heavily, run at a low temperature to recover perhaps 60 per cent of the zinc content of the ore as spelter, and the retort residues, containing the remaining 40 per cent of the zinc, burned on Wetherill grates for the recovery of the rest of the zinc as zinc oxide. The retort furnace and Wetherill grate would thus supplement each other, each performing that part of the work for which it is best suited. This scheme has been tried by several plants in recent years, but has not proved to be practicable.

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[24] Hofman, H. O., Metallurgy of zinc and cadmium. 1922, p. 301.

[24a] Some operators disagree with this statement; their experience has been that the blanketing and slow-burning action of the residue interferes with proper recovery from a mixture of residues and ore.

Various modifications of the method of burning residues for the production of oxide and of apparatus for carrying it out have been suggested by Stolzenwald,[25] Hommel,[26] Köhler and Baltin,[27] Uehlendahl,[28] and Eichhorn.[29]

MISCELLANEOUS PROCESSES

Various methods other than those already described have been suggested for the treatment of retort residues. Most of these are schemes for leaching with acids or with ammonia. MacIvor proposed[30] to mix the residues with sulphuric acid, knead the mixture into a pasty mass, and heat to a temperature of 300 to 400° C., to convert the zinc to zinc sulphate and to dehydrate gelatinous silica. The zinc was then to be leached out with water. A modification of this[31] consisted of a direct leach with sulphuric acid and calcium chloride, whereby the zinc was dissolved as zinc chloride. Boguski patented a process[32] for leaching with ammonia.

TREATMENT OF BROKEN RETORTS AND CONDENSERS

Old retorts contain 6 to 15 per cent zinc, mainly in the form of zinc spinel (zinc aluminate) and zinc silicate, though there is usually some metallic zinc in crusts near the front ends of the retorts. Formerly broken retorts were cleaned of adhering slag and crusts and then crushed for use as burnt clay for new retorts. At present it is more common to use only new clay for the retorts, and to use the crushed old retorts in the mixture for condensers.

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[25] Stolzenwald, Gustav, Process of treating zinc-retort residues, United States patent 881,355, Mar. 10, 1908; Method realizing the value of material containing zinc and simultaneously profiting by the zinc and carbon constituents contained in the residue of zinc works, British patent 18,134, Nov. 8, 1906; Process for the treatment of zinciferous materials and simultaneously recovering the zinc and carbon contained in the residues from metallurgical treatment of zinc, French patent 368,458, Oct. 6, 1906; same as British patent, German patent 180,981, Feb. 9, 1907.

[26] Hommel, Waldemar, Improvements in or relating to the extraction of zinc from ores and zinc residues, British patent 27,020, Mar. 9, 1909; same title, French patent 401,557, July 30, 1909.

[27] Köhler, R., and Baltin, O., Process for utilizing the residues obtained in the distillation of zinc, United States patent 1,275,045, Aug. 6, 1918.

[28] Uehlendahl, Otto, Process and furnace for obtaining zinc from residues, slags, and low-grade ores, German patent 271,826, Mar. 21, 1914.

[29] Eichhorn, Konrad, Process for working up zinc-retort residues, German patent 309,162, Oct. 23, 1919.

[30] MacIvor, R. W. E., Improvements in or relating to the treatment of zinc residues, British patent 17,001, July 25, 1907.

[31] MacIvor, R. W. E., Improvements in or relating to the treatment of zinc residues, British patent 21,440, Sept. 26, 1907.

[32] Boguski, J. J. [Process for recovering from calamine, especially from ores of low zinc content, and also from spelter residues, by leaching with ammonia], German patent 271,658, Mar. 17, 1914.

Discarded condensers always have much zinc attached to them as metal and oxide crusts. They may be crushed and added to the retort charge, or they may be cleaned of metal-bearing crusts and only the richest parts broken off and recharged. By the use of jigs or tables the metal content of condensers can be recovered as a rich concentrate. At several plants a hand jig or a simple power jig is used for this work.

At one smelter where a hand jig is used 12,000 pounds of broken condensers and retort ends are concentrated per day, with the production of 4,000 pounds of concentrate containing 65 per cent zinc. This concentrate is saved until enough is on hand to be treated in a furnace block by itself, as the zinc distills from it at a lower temperature than from ore.

At another smelter the best parts of the condensers are crushed and charged to the retorts; the rest of the condensers and the best parts of the old retorts are jigged, making a concentrate that assays 65 per cent zinc and a tailing that assays 1 to 2 per cent zinc.

At a third smelter a more elaborate plant for concentrating retorts and condensers is operated in the same building with a concentrating plant for residues. The retorts and condensers are milled separately on alternate days. They are crushed and screened; the fines go to James tables and the oversize to a Richards jig. The concentrates contain about 80 per cent zinc. The tailings, consisting of clean burnt fireclay, carry between 1 and 2 per cent zinc. The table tailings are discarded. The jig tailings from the concentration of the retorts are used for making new retorts; the batch for these is made up of 25 parts new burnt clay, 25 parts reclaimed clay from old retorts, and 50 parts raw clay. Similarly the clay recovered from the condensers is used for new condensers, but for these no new burnt clay is used; it is replaced entirely by the reclaimed clay.

At many zinc smelters that at present have no concentrating plants the condensers are being saved with a view to concentrating them at some future time.

SUMMARY

The author has described a variety of methods for the treatment of zinc retort residues. Most of these methods are or have been in actual use in one or more zinc smelters. Different conditions require different methods; some methods have been successful in certain plants and unsuccessful in others.

It is probable that conditions at some plants prohibit the profitable treatment of retort residues by any method; on the other hand, it is probable that many companies which do not now treat their residues could do so by selecting the proper method and adapting it to their particular conditions.

The treatment of old condensers is simpler than that of retort residues, and it seems that most zinc smelters could with profit build at least a simple jigging plant for their concentration.

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